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THE CANMET FERRIC CHLORIDE LEACH PROCESS - SOLIDS RETENTION TIME AND ACID CONSUMPTION IN LEACHING

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THE CANMET FERRIC CHLORIDE LEACH PROCESS - SOLIDS RETENTION TIME AND ACID CONSUMPTION IN LEACHING

by

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ABSTRACT

CANMET has developed a $FeCl_3$ leach Process (FCL) to treat complex fine-grained base metal sulphide ores. Batch leaching of a pyritic sulphide concentrate with $FeCl_3$ was investigated with respect to retention time and HCl concentration. A two stage leach of a pyritic bulk concentrate using lixiviants containing 0.90 M $FeCl_3$, 3.0 M NaCl and 0.1 M HCl extracted in 6.0 h at 107-108°C, 99.6% Zn, 97.0% Ag, 99.5% Pb and 98.1% Cu. The 1.5 h first stage leach yielded a pH 0.5 pregnant solution and consumed 23.5 kg HCl/tonne concentrate. The 4.5 h second leach stage did not consume any acid. At present, the two stage FCL Process requires operation in a continuous pilot plant to evaluate leaching and to study the effect of impurity build up with recycle.

Keywords: $FeCl_3$, pyritic bulk concentrate, two stage

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INTRODUCTION

Large deposits of complex fine-grained base metal sulphide ores occur in many locations world-wide. Although many of these deposits are relatively high grade, their economic exploitability has been severely hindered by the difficulty in achieving high metal recoveries while producing readily saleable selective flotation concentrates.

Metal recoveries from such ores can be significantly improved by producing a bulk concentrate rather than selective concentrates. Previous CANMET studies have demonstrated that bulk flotation of New Brunswick ores can increase recovery by at least 25 % of ore value (1, 2). Bulk concentrates with grades of 30 % to 40% combined metals and recoveries in excess of 90%, are possible. These bulk concentrates, because of their complexity and relatively low grade, are not amenable to treatement by conventional extraction/refining technology.

CANMET has developed the Ferric Chloride Leach (FCL) Process to treat complex, low grade, sulphide bulk concentrates (3, 4, 5, 6, 7). The FCL process flowsheet for application to a Zn, Pb, Ag, Cu Bulk Concentrate is shown in Figure 1. The flowsheet consists of seven circuits; a leach circuit, three product recovery circuits, an iron rejection oxyhydrolysis circuit, an effluent treatment circuit, and a lixiviant regeneration circuit. The process provides very high recoveries of lead, zinc, copper and silver and is relatively insensitive to concentrate grade.

The reactive metal sulphides are converted to soluble metal chlorides and elemental sulphur using dissolved ferric iron:

$$ZnS + 2FeCl_3 \rightarrow ZnCl_2 + 2FeCl_2 + S^\circ \quad (1)$$

$$PbS + 2FeCl_3 \rightarrow PbCl_2 + FeCl_2 + S^\circ \quad (2)$$

$$CuFeS_2 + 3FeCl_3 \rightarrow CuCl + 4FeCl_2 + 2S^\circ \quad (3)$$

$$FeS + 2FeCl_3 \rightarrow 3FeCl_2 + S^\circ \qquad (4)$$

The metal chlorides are treated to produce refined zinc and lead and a high grade copper/silver cement. Where economics warrant, the sulphur can be recovered. The process maximizes lixiviant recycle and the iron bearing residue is relatively free of heavy metal impurities. Since no SO_2 is produced, a H_2SO_4 recovery circuit is not required.

To maximize metal recovery and to minimize operating costs and process control problems, a two stage leach with inter-stage thickening is recommended (5). The first stage leach pregnant is fed to the metal recovery circuits whereas the second stage leach pregnant is regenerated by chlorination and recycled to the first stage leach. Sulphate build-up in the leach circuit is controlled by $CaCl_2$ addition to precipitate $CaSO_4$. The build-up of impurities is controlled by bleed steam treatment. A surfactant is added to both leach stages to maintain porous sulphur coatings around the sulphide particles thereby permitting enhanced metal extraction (8).

The first stage leach has the following characteristics:

1. The leach is operated under reducing conditions, where the volume and concentration of Fe^{3+} is sub-stoichiometric with respect to the quantity of metal sulphides present in the concentrate. This condition ensures that the stage one leach pregnant contains a maximum of reduced species (Fe^{2+} and Cu^{1+}) and so contains a minimum of Fe^{3+}.

2. The lixiviant contains sufficient acid to neutralize the acid consumers in the concentrate and to prevent iron hydrolysis and precipitation in the leach and product recovery circuits.

The second stage leach has the following characteristics:

1. The leach is operated under oxidizing conditions, to maximize metal extractions, maintaining a minimum 25% stoichiometric excess of Fe^{3+}.

2. The leach does not consume acid. Therefore the stage two lixiviant need only contain sufficient acidity to prevent any precipitation of metals.

Previous studies at CANMET have demonstrated that the FCL process is technically feasible (9). Testing at pilot scale indicated that 99.4 % Zn, 99.6 % Pb, 97.8 % Cu and 97.0 % Ag extractions were achieved in < 8 h at initial acid concentrations of 0.31 to 0.59 M HCl. The acid consumption during leaching was excessively high at 44 kg/tonne concentrate. Additional bench scale tests achieved 99.3 % Zn, 99.7 % Pb, 97.4 % Cu and 95.8 % Ag extractions in 7.5 h at initial acid concentrations of 0.22 M HCl (10). The reduction in acid was in part made possible by a process change from lead to iron cementation of Cu/Ag (11). The acid consumption was reduced to about 30 kg/tonne.

The objectives of this present study are to reduce the acid consumption while maintaining the high metal recoveries above.

EXPERIMENTAL

A bulk concentrate prepared from Brunswick Mining and Smelting ore was used in this present study . The mineralogy of the bulk concentrate has been extensively studied and reported previously (12, 13). Possible acid consumers were identified and their respective concentrations were calculated ($FeCO_3$ 1.8; $CaCO_3$ 0.54; $MgCO_3$ 0.38 weight percent). Assuming that each carbonate reacts with HCl as below and that each reaction goes to completion, the minimum HCl consumption would be 20.8 kg HCl/tonne concentrate.

$$FeCO_3 + 2HCl \rightarrow FeCl_2 + CO_2 + H_2O \qquad (5)$$

$$CaCO_3 + 2HCl \rightarrow CaCl_2 + CO_2 + H_2O \quad (6)$$

$$MgCO_3 + 2HCl \rightarrow MgCl_2 + CO_2 + H_2O \quad (7)$$

Table 1 and Figure 2 show the chemical and particle size analysis for the concentrate used for the $FeCl_3$ leaching tests. The bulk concentrate is very finely sized. The mean particle size was 14.6 μM with a specific surface area of 0.634 m^2/cm^3. The concentrate was 81.9 vol% finer than 44μm (325 mesh). By comparison, a previous determination using wet screening gave a result of 86.9 wt% finer than 44μm (5).

Table 1

Chemical Analyses of the Bulk Concentrate, wt%

Zn	Pb	Cu	Ag[1]	Fe	S (Total)	S°	SO_4/S	As	Sb	Cd[1]	Bi[1]
28.8	8.5	0.64	251.8	18.7	34.27	0.86	3.11	0.4	0.1	470	350

(1) ppm

To approximate the circulating load of metal values as anticipated in two-stage leaching (5), lixiviants for stage 1 and 2 were prepared as shown in Table 2. Reagent grade chemicals were used to prepare all test solutions.

Leach tests were carried out in 2.0 L round bottom glass reaction vessels. Each vessel was agitated at a constant speed of 600 RPM using a two-bladed polypropylene agitator. The leach temperature was controlled to $\pm 2°C$ and a water-cooled condenser was used to minimize evaporative losses.

At the start of each test, 1.25 L of $FeCl_3$ Lixiviant 1, $CaCl_2.2H_2O$, and the surfactant were added to the reaction vessel and heated to 90°C under agitation and N_2 atmosphere. To reach the desired temperature, small portions of weighed concentrate were added slowly over a 5 min period. The leach reactions are exothermic. Once the desired temperature was reached, t = 0.0 hour was defined, and the leaching was continued for the specified time.

Table 2

Lixiviant Composition

Ion	Stage 1 Lixiviant (g/L)	Stage 2 Lixiviant (g/L)
Fe^{3+}	50	50
Na^+	70	70
Pb^{2+}	6	6
Ca^{2+}	3	3
$(SO_4)^{2-}$	3	4.5
Zn^{2+}	20	4
Cu^+	0.5	0.0
Ag^+	0.015	0.0

At the end of the test, the hot leach slurry was immediately vacuum filtered on a Buchner filter. A sample of pregnant solution was stored in a sealed bottle to prevent the oxidation of Fe^{2+} iron prior to analyses. A 10 g grab sample was obtained from the moist filter cake and repulped in 50 mL of 80-90°C 1 M NaCl, followed by displacement washes of 2 x 50 mL 80-90°C distilled water. The washed stage one leach residue was dried overnight at 85°C, weighed and submitted for assay. The bulk of the stage one leach residue was retained for stage two leaching.

The stage two leach was conducted as above using 1.25 L of $FeCl_3$ lixiviant 2. An air atmosphere was maintained above the slurry. The second stage leach slurry was filtered as above and the residue was repulped with 200 mL of 80-90°C 1 M NaCl, followed by displacement washing of 2 x 200 mL 80-90°C distilled water. The stage two residue was dried overnight at 85°C, weighed and assayed.

Solution pH and emf measurements were made using combination electrodes of the double junction Ag/AgCl reference type rated at 135°C. The measurements were made both hot and at room temperature.

The determination of free HCl was performed by complexation titrations using CTDA (cyclohexylenedinitrilo-tetraacetic acid) as the complexing agent and standardized NaOH titrant. Sample aliquots were limited to 50 mg Fe and titrated automatically to the first derivative end point. Total Fe and Fe^{2+} concentrations were determined by potentiometric titration using standard $K_2Cr_2O_7$ solution. The soluble Cu, Zn, Ag, As, and Ca concentrations were determined by atomic absorption spectrophotometry (AAS) and the soluble sulphate was determined gravimetrically. Solid samples were analyzed by fire assay techniques with AAS finish. The fixed conditions for the leach tests are given in Table 3.

Table 3

Fixed Leach Conditions

Parameter	Stage 1	Stage 2
Feed	202.5 g bulk conc.	Stage 1 leach residue
Lixiviant	1.25 L Lix 1	1.25 L Lix 2
Surfactant	250 g/tonne	250 g/tonne
$CaCl_2.2H_2O$	49.4 kg/tonne	-
Temperature	107 - 108°C	107 - 108°C
HCl	0.07 or 0.10 M	0.07 or 0.10 M
Atmosphere	N_2	Air

RESULTS AND DISCUSSIONS

STAGE ONE

Retention Time: 0.07 M HCl

At the lower acid concentration, the data suggest that Zn, Cu, Ag and Fe extractions decreased with increased retention time > 2 h. The Pb extraction reached a maximum after 1.0 h and remained constant to 3 h (Table 4). The Cu extraction decreased from about 66% to 9 % between 2 and 3 h (Figure 3). During the same time period, the corresponding Fe extraction decreased from 23% to 10% Fe.

The data suggest that Fe hydrolysis and co-precipitation of Cu with Fe took place but this was not directly verified. The literature indicates that Cu entrainment with goethite (FeOOH) is possible via cationic sorption (14).

The elimination of trace elements (As, Sb, Cd, Mg, Mn, Bi) with Fe, as stable compounds would be advantageous and would reduce the demands on the waste rejection circuits. The leaching of As, Sb and Cd at 0.07 M HCl was similar to previous studies at 0.43 M HCl (2). As, Sb and Cd all leached rapidly with first stage extraction of 60%, 65% and 70%, respectively, after 1.5 h retention time. No decrease in trace element extraction was observed with increased retention time to 3 h.

Retention Time: 0.10 M HCl

The kinetic leach data show that significant metal dissolution occured in only 1 h (Table 5). The Zn, Cu, and Fe extractions increased marginally between 1 and 3 h retention time (Figure 4). The Pb and Ag extractions do not appear to change between 1 and 3 h retention. The decreased in leaching rates after 1 h are believed due to a large decrease in the Fe^{3+}concentration. It is not known if the increased Fe extraction was due to pyrite dissolution or simply due to the

leaching of Fe values associated with chalcopyrite and marmatite. Fe is known to be present in solid solution with the Zn as the mineral marmatite (3).

Fe^{3+} Utilization: 0.10 M HCl

The rapid leach kinetics of Zn and Pb resulted in a rapid reduction of Fe^{3+} to Fe^{2+}. As shown in Figure 5, the slurry redox potential decreased from +650 to +260 mV (Ag/AgCl/Pt) after 30 min retention time. The slurry redox potential remained at $+260 \pm 20$ mV for the duration of the 3.0 h leach. The 3.0 h leach pregnant solution contained 420 ppm Fe^{3+}. The decrease in Fe^{3+} concentration as indicated by the rapid decrease in slurry redox potential is believed to be the principal reason for the rapid decrease in metal extraction. HCl does not contribute significantly to the leaching of metal sulphides in a $FeCl_3$ leach system. The pregnant solution pH were generally at pH 0.5.

Table 4

Retention Time Stage One Extraction 0.07 M HCl

Retention Time	Pregnant			Metal Extraction (%)[1]				
	Fe^{3+}	emf	pH					
(h)	(mg/L)	(+mV)		Pb	Zn	Cu	Ag	Fe
1.0	1815	267	1.5	97.8	65.8	58.9	65.9	33.8
1.0[2]	1500	255	1.4	97.0	71.3	66.2	65.9	24.2
1.5	4130	260	1.5	97.2	64.0	45.7	66.3	26.8
2.0	2570	250	1.6	98.0	67.0	65.3	73.9	23.0
3.0	1500	260	1.7	97.8	62.0	10.3	62.9	10.9

(1) Metal extractions based on residue assays.

(2) Duplicate run.

Table 5

Retention Time Stage One Extraction 0.10 M HCl

Retention Time	Pregnant			Metal Extraction (%) [1]				
	Fe^{3+}	emf	pH					
(h)	(mg/L)	(+mV)		Pb	Zn	Cu	Ag	Fe
1.0	1470	260	1.1	99.1	62.9	61.4	55.7	26.0
1.5	640	265	0.54	-	58.3	52.9	-	14.4
2.0	1100	_	0.46	98.0	66.4	61.1	-	30.2
3.0	420	255	0.50	98.9	64.8	64.8	55.9	29.0

(1) Metal extraction based on residue assay

<u>Fe^{3+} Utilization: 0.07 M HCl</u>

At the lower acid concentration, the metal leach kinetics were initially as fast as at the higher acid concentration. The metal extraction are shown in Table 4. The rapid depletion of Fe^{3+} decreased the emf from +650 to + 267 mV in 1.0 h (Figure 5). After 1.0 h the emf stabilized at + 260 mV, while the pH and Fe_{3+} concentrations generally increased with increased retention time. At 0.07 M HCl, the leach pregnants contained high Fe^{3+} concentrations, typically > 1200 mg/L. The difference in Fe^{3+} concentrations suggests that a 0.07 M HCl system are more prone to Fe^{2+} oxidation than a 0.1 M HCl system. The mechanism of Fe^{2+} oxidation is not known at this time. Since the leach was conducted under a N_2 atmoshere it is unlikely that O_2 was the oxidant, although the increase in pH does support the oxidation as below:

$$2Fe^{2+} + 2H^{+} + 0.5\ O_2 \rightarrow 2Fe_3 + H_2O \qquad (8)$$

Further complicating the leach environment is the redox reaction between soluble copper and iron:

$$Fe^{3+} + Cu^{+} \rightarrow Fe^{2} + Cu^{2+} \qquad (9)$$

The solution emf at < 300 mV, was not a reliable guide to the concentration of Fe^{3+} in solution because of the complication presented by other reactions which also influence the redox potential.

HCl Utilization: Stage One

The variation in leach pregnant pH with retention time was as shown in Tables 1 and 2 and in Figure 5. The 0.1 M HCl leach was stable and reproducible yielding a 0.5 pH pregnant after 1.5 to 3.0 h retention. From previous work, leach pregnants with a pH in the range 0.5 to 1.0 were found to be acceptable for treatment in the post leach product recovery circuits (10). A 0.07 M HCl leach gave increased solution pH with increased retention. After 3.0 h, a pH of 1.7 was obtained which is too high as post leach Fe precipitation may occur. The HCl consumptions were equivalent to 15.2 and 23.5 kg HCl/tonne at initial HCl concentrations of 0.07 and 0.1 M HCl, respectively and correspond to 0.73 and 1.13 times the stoichiometric HCl consumption.

Stage One: Summary

The leach conducted at 0.1M HCl gave the more acceptable results. The 0.1M HCl leach consumed more acid than the leach conducted at 0.07M HCl (23.5 vs 15.2 kg HCl/tonne), but the 0.1 M lixiviant produced a stable pH 0.5 pregnant solution and no Cu losses were experienced with retention time <3.0 h. The target specification for Fe^{3+} in the stage one pregnant solution was arbritraily set at <500 mg/L (5). The <500 mg/L Fe^{3+} specification was met by extending (the 0.1 M HCl leach) retention time to 3.0 h.

The stage one leach at 0.1M could be operated at 1.0 h retention time with no deleterious effect on metal extraction, which suggests a short retention time and the consideration of strategies other than prolonged retention to control the Fe^{3+} concentration. An option worth consideration would be a split addition of fresh concentrate to the stage one leach. This approach has been used previously (7) and should be studied by adding both fresh concentrate and FCL Process recycle solids to the 1.0 h leach slurry. It is expected that an additional 0.5 h retention will reduce the Fe^{3+} concentration to < 1000 mg/L.

STAGE TWO

Retention Time

Stage two leach tests were carried out using stage one residues as feed and Lixiviant 2 containing 0.07 and 0.1 M HCl. The stage two leach retention time was varied between 2.0 and 4.5 h. At 0.07 M HCl the incremental copper extraction increased significantly with time. This was not the case at 0.10 M HCl. The incremental extraction of the other metals were little effected by extraction time. Average incremental metal extractions in stage two at 0.07 and 0.10 M HCl were: Pb 2.1%, Zn 32.5%, Ag 28.2%, Fe 10.3% and Pb 0.9%, Zn 36.1%, Cu 37.4%, Ag 40.6%, Fe 11.1%, respectively. The incremental extractions produced in stage two were as shown in Tables 6 and 7 while Figures 6 and 7 show the overall extraction for stages one and two combined. The minimum acceptable Zn, Ag, Pb and Cu extractions of 95% were achieved at 4.0 hours cumulative retention time. Small increases in Zn, Cu and Ag extractions were obtained at increased retention time.

Table 6

Stage Two $FeCl_3$ Leaching Incremental Extractions 0.07 M HCl

Retention Time (h)	Metal Extraction (%)[1]				
	Pb	Zn	Cu	Ag	Fe
2.0	2.5	24.8	28.2	27.8	14.0
2.5	na	na	na	21.4	na
3.0	1.7	32.4	38.5	29.2	0.0
4.5	2.4	35.3	52.3	30.0	5.4
4.5[2]	1.7	37.6	88.1	33.0	21.8

(1) Metal extraction based on final residue analysis.
(2) Duplicate run.

The empirical relationships between Zn extraction and retention time were derived from Figures 6 and 7. To achieve 99 % Zn extraction the cumulative retention time of 5.2 and 4.4 h, respectively, would be required for the 0.07 and 0.10 M HCl leaches.

Table 7

Stage Two $FeCl_3$ Leaching Incremental Extractions 0.10 M HCl

Retention Time (h)	Metal Extraction (%)[1]				
	Pb	Zn	Cu	Ag	Fe
3.0	0.6	35.7	34.8	40.4	12.4
3.0[2]	1.6	32.9	37.1	na	5.0
4.5	na	41.3	45.2	na	21.6
4.5[2]	0.6	34.5	32.6	40.8	5.4

(1) Metal extractions based on final residue analysis.
(2) Duplicate run.

Copper extraction behaved differently at 0.07 than at 0.1 M HCl. This difference was, however, expected as severe Cu precipitation occurred during the stage one leach (0.07 M)

which may have given rise to different Cu minerals and compounds for subsequent leaching in stage two. At 0.07 M HCl, the stage two Cu leach data were interpreted to increase logarithmically to achieve a extrapolated 98 % extraction in 6.0 h. The logarithmic Cu leach curve may correspond to the initial rapid leaching of less refractory Cu minerals, followed by the slower leaching of refractory chalcopyrite Cu. At 0.1 M HCl the stage two Cu extraction data gave a linear relationship versus retention time and a calculated Cu extraction of 96.6% after 5.2 h.

The behaviour of Ag at 0.07 M and 0.1 M HCl was interpreted as a logarithmic and a linear dependence respectively with increased retention time. That is, Ag behaved as Cu.

In stage two, arsenopyrite and Sb-bearing tetrahedrite were further leached to produce soluble arsenic, antimony and silver values. Incremental metal extraction in stage two were 4 % As, 19 % Sb, 26 % Cd, and 30 % Ag (0.07 M HCl, 4.5 h stage two). The final 6.0 h extractions were 64 % As, 84 % Sb, 96 % Cd and 96.3 % Ag. With increased retention to 7.5 h, the As extraction increased to 76% and the Sb and Cd extractions remained constant at 84 % and 96 %, respectively. Clearly, the 6.0-hour retention time offers process advantages with reduced As impurity leaching with acceptable Zn, Ag, Pb and Cu extraction.

At 6.0 h retention time, the predicted metal extractions derived graphically were as below:

	Pb	Zn	Cu	Ag
0.07 M HCl	99.6	99.4	98.0	95.8
0.1 M HCl	99.5	99.3	97.7	96.8

The two stage metal extractions reported herein were based on the final residue and head assays. Consideration was given to the loss of metal values contained in the stage one residue sample. This sample (which contained 2.4, 2.4, 2.1 and 0.04 wt% of Zn Cu, Ag and Pb contained in the concentrate) was assumed in stage two, to leach as did the final residue. This assumption was based on excess Fe^{3+} in the stage two leach pregnant values which can quantitatively

solubilize more than 6 wt% of the metal values contained on the bulk concentrate feed (6.8 g/L of Fe^{3+}iron remaining in the 4.5 h second stage leach slurry Table 7.

HCl Utilization

During the stage two leach the HCl concentration increased, on average by 0.097 and 0.11 M HCl at initial HCl concentrations of 0.07 and 0.1 M (Table 8). This phenomena has been observed previously in the $FeCl_3$ leaching of pyritic concentrates (9). The HCl concentrations generally increased with increased stage two retention time in the presence of decreasing Fe^{3+} concentrations. These facts suggest that the increased HCl concentrations were possibly associated with Fe^{3+} oxidation followed by Fe^{3+} hydrolysis and precipitation.

$$2\ Fe^{2+} + 2\ H^{+} + 0.5 O_2 \rightarrow 2 Fe^{3+} + H_2O \qquad (10)$$

$$Fe^{3+} + 4H_2O \rightarrow FeOOH + 3H^{+} \qquad (11)$$

For each mole of Fe^{2+} precipitated as FeOOH, a net generation of 2 moles H^{1+} takes place. In practice, a net acid generation (average) of 0.102 M is equivalent to 3.6 g Fe^{2+} precipitated as FeOOH. X-ray diffraction studies, however, could not identify the presence of basic iron compounds in the predominately FeS_2 stage two leach residue. The process advantages of in-leach precipitation of Fe and simultaneous generation of acid are considerable. Both the capital and operating costs of a FCL plant would be reduced. The acid fortified stage two pregnant containing nominally 0.2 M HCl, would be recycled to the stage one leach. In previous $FeCl_3$ tests at 0.22 M HCl, acceptable two stage metal extraction were obtained and the stage one leach pregnant contained 0.07 M HCl and no Fe^{3+} iron (10).

Table 8

Stage Two $FeCl_3$ Leaching HCl Behaviour

HCl Initial - 0.07 M HCl			HCl Initial - 0.10 M HCl		
Retention Time	Pregnant[1]		Retention Time	Pregnant[1]	
	Fe^{3+}	HCl		Fe^{3+}	HCl
(h)	(g/L)	(M)	(h)	(g/L)	(M)
2.0	13.6	0.15	3.0	9.0	0.19
2.5	10.7	0.17	3.0	9.5	0.20
3.0	9.9	0.18	3.0	10.5	0.25
4.5	8.0	0.13	3.0	11.0	0.18
4.5	8.2	0.18	4.5	7.5	0.21
4.5	6.8	0.19	4.5	8.0	0.22

(1) concentrations adjusted for 1.25 L

CONCLUSIONS AND RECOMMENDATIONS

Batch leaching of a pyritic sulphide concentrate with $FeCl_3$ was investigated with respect to retention time and HCl concentration. Under all conditions tested very high extractions of Zn, Pb, Cu and Ag were achieved in less than 6.0 h. The two stage leach consumed acid in stage one and generated acid in stage two. The leach may not require the addition of purchased acid to neutralize the acid consumers present in the New Brunswick bulk concentrate. The preferred two stage leach conditions were as below with 6.0 h overall retention time.

Stage One: Feed - Bulk concentrate 202.5 g
Lixiviant - 1.25 L containing 3.0 M NaCl, 0.90 M $FeCl_3$, 0.1 M HCl
Additives - 250 g surfactant/tonne concentrate
Temperature - 107-108°C
Retention Time - 1.5 h

Stage Two: Feed - stage one residue
Lixiviant - 1.25 L containing 3.0 M NaCl, 0.90 M $FeCl_3$, 0.1 M HCl
Additives - 250 g surfactant/tonne concentrate
Temperature - 107-108°C
Retention Time - 4.5 h

The observed metal extractions under these conditions were Pb, 99.6% Zn, 97.0% Ag, 99.5% Pb 98.1 Cu and 36% Fe.

After 1.5 h retention in stage one, the HCl consumption was 23.5 kg HCl/tonne concentrate corresponding to 1.13 times the stoichiometric HCl required to neutralize the carbonates. The 1.5 h leach pregnant contained about 1500 mg/L Fe^{3+} at pH 0.5.

In stage two, HCl was not consumed but increased from 0.1 to about 0.2 M in the pregnant solution. The increase in the HCl concentration is believed to be associated with Fe precipitation. The additional acid values could be advantageously utilized in the state one leach. In previous work, higher HCl concentrations promoted lower Fe^{3+} concentrations in the stage one leach pregnant.

The $FeCl_3$ leaching of pyritic bulk concentrates achieved high metal extractions and should be further developed. It is recommended that the preferred leach conditions be verified in a continuous pilot plant using recycled process liquors. The continuous mode, leach operation, will facilitate the study of impurity build up and will allow for equilibrated leach liquors to be evaluated for compatability with other unit operations comprizing the CANMET FCL Process.

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APPENDIX

Minor Element Metal Extraction Results, 0.07 M HCl

Conditions	As (%)	Sb (%)	Cd (%)	Bi (%)
Stage One, 1.5 h	49.2	72.9	69.8	69.1
Stage Two, 4.5 h	26.5	13.2	26.3	69.7
Overall, 6 h	75.7	86.1	96.1	78.8

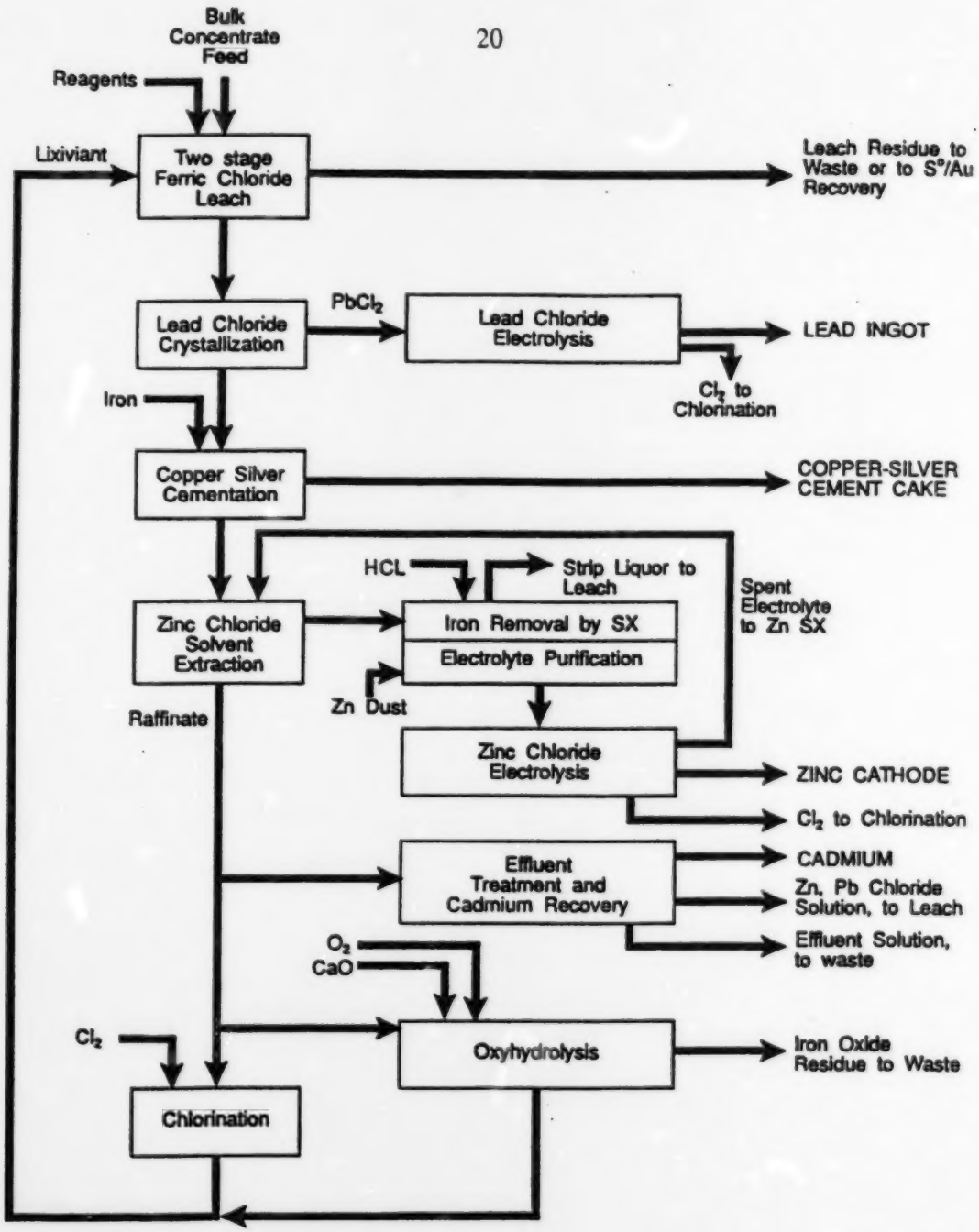

Figure 1 CANMET FCL Process Flow Diagram

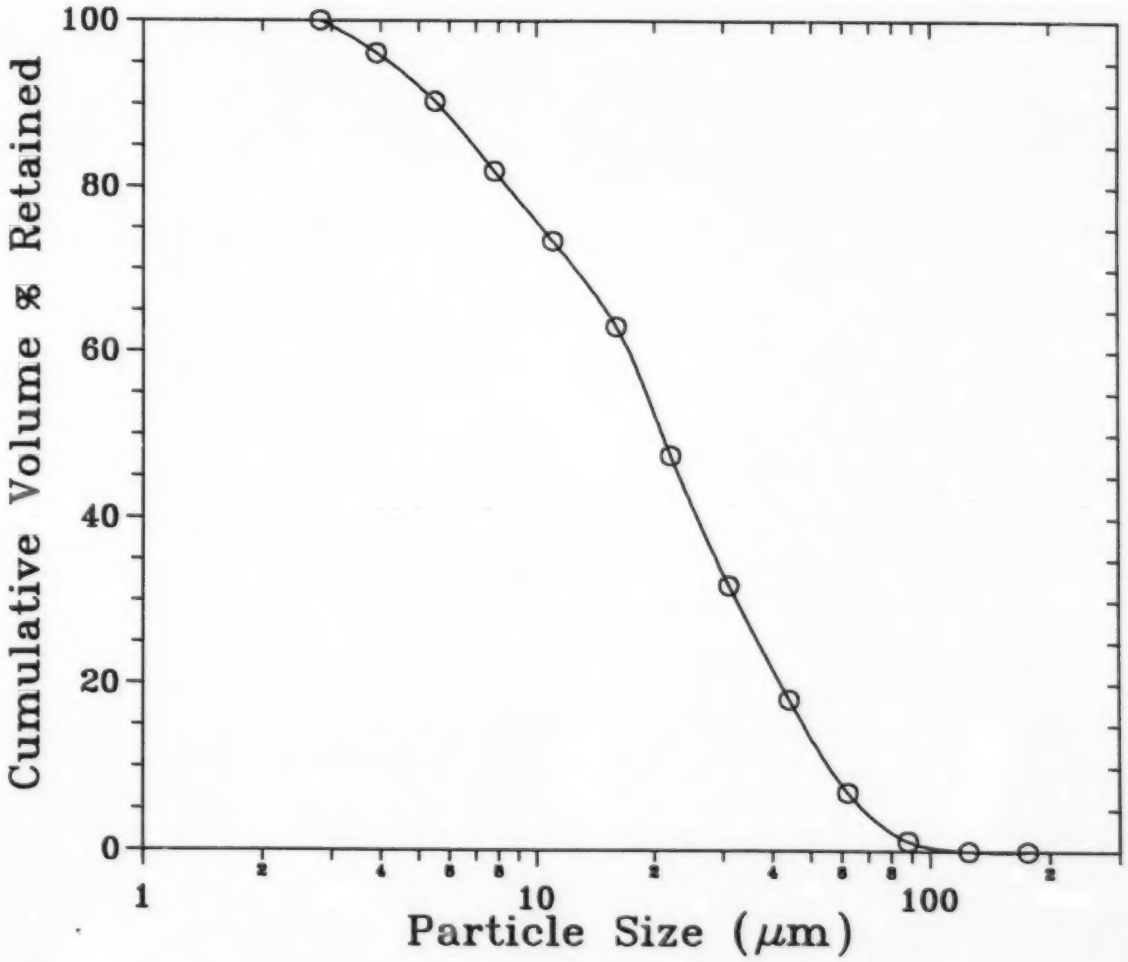

Figure 2 Bulk Concentrate Particle Size Analysis (Wet technique - Leeds and Northrup Laser Particle Size Analyzer)

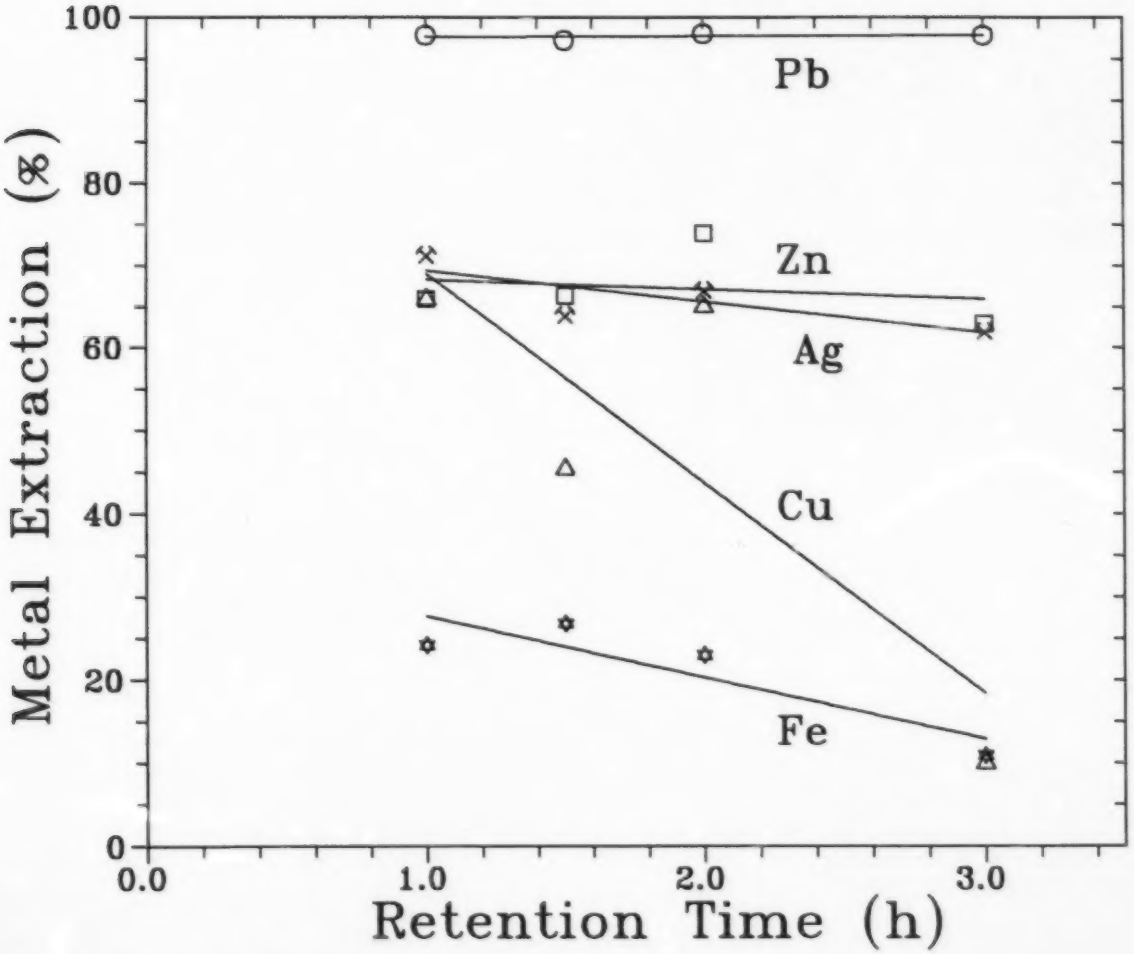

Figure 3 Stage 1 Metal Extraction (0.07M HCl)

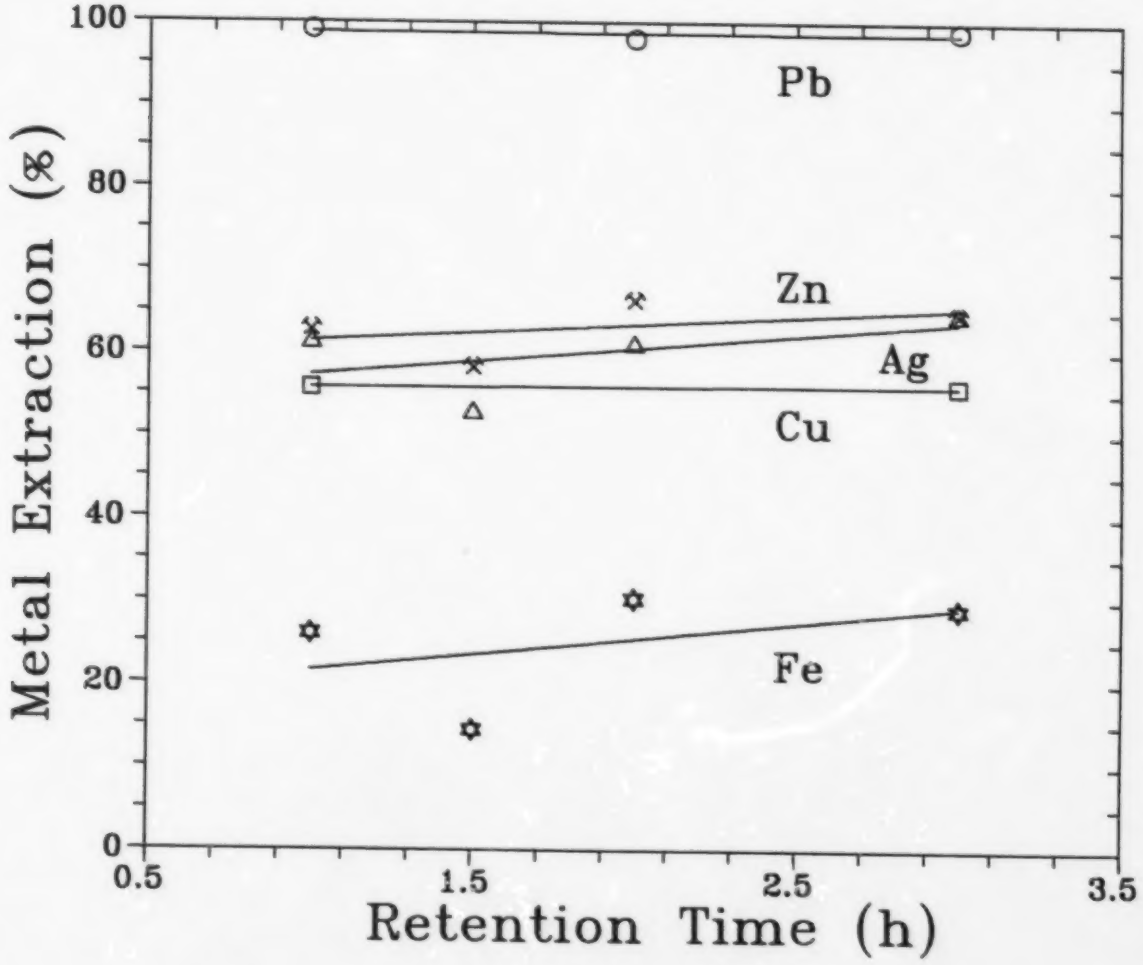

Figure 4 Stage 1 Metal Extraction (0.10M HCl)

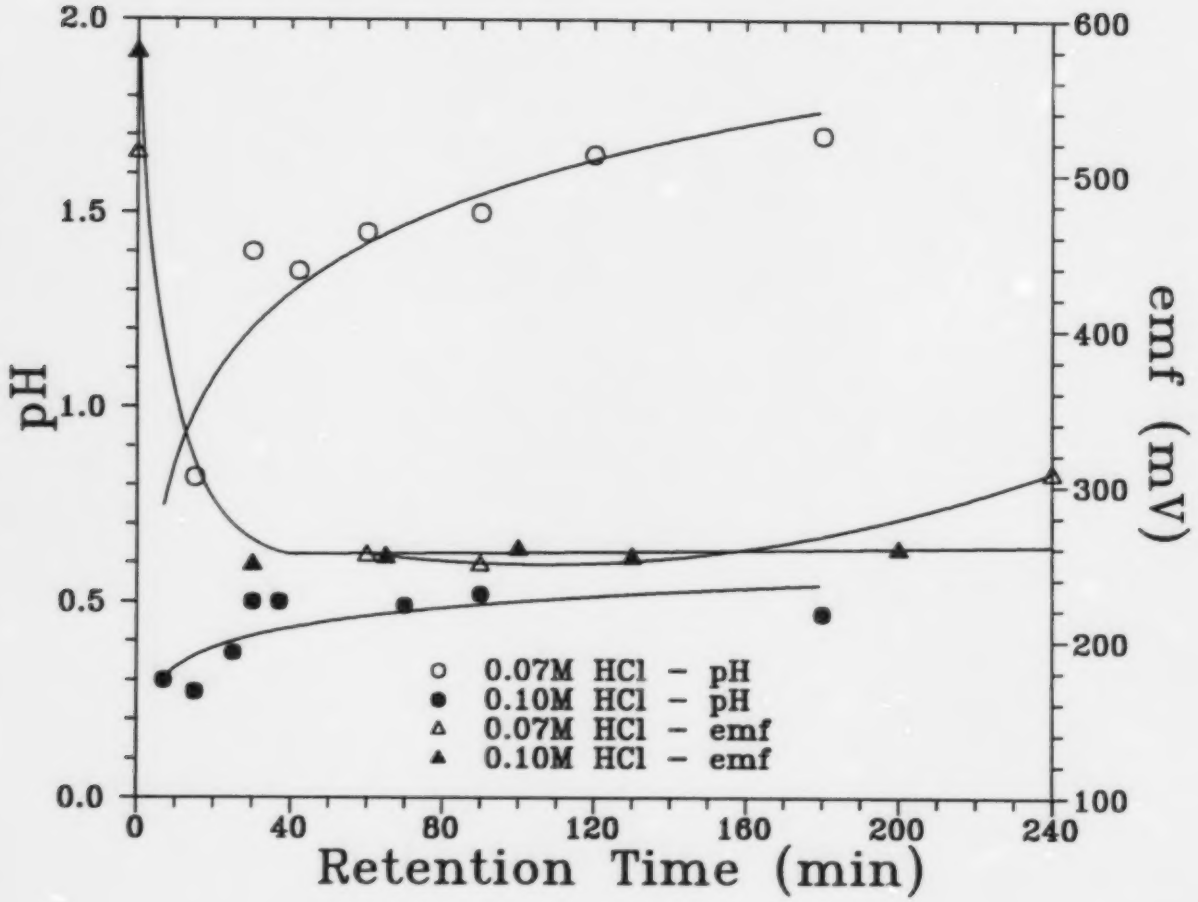

Figure 5 Stage 1 Leach Liquor Characteristics

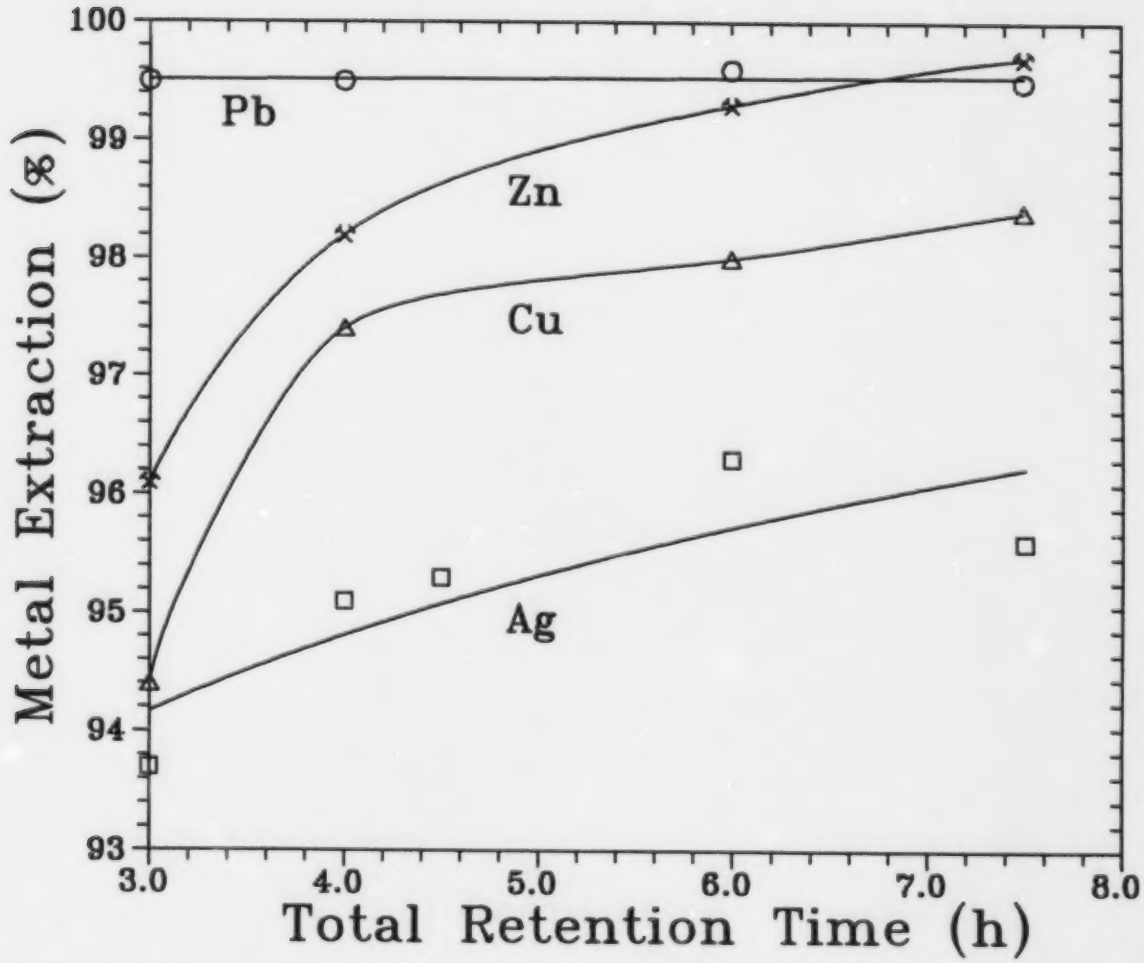

Figure 6 Cumulative Stage 1 and 2 Metal Extraction (0.07M HCl)

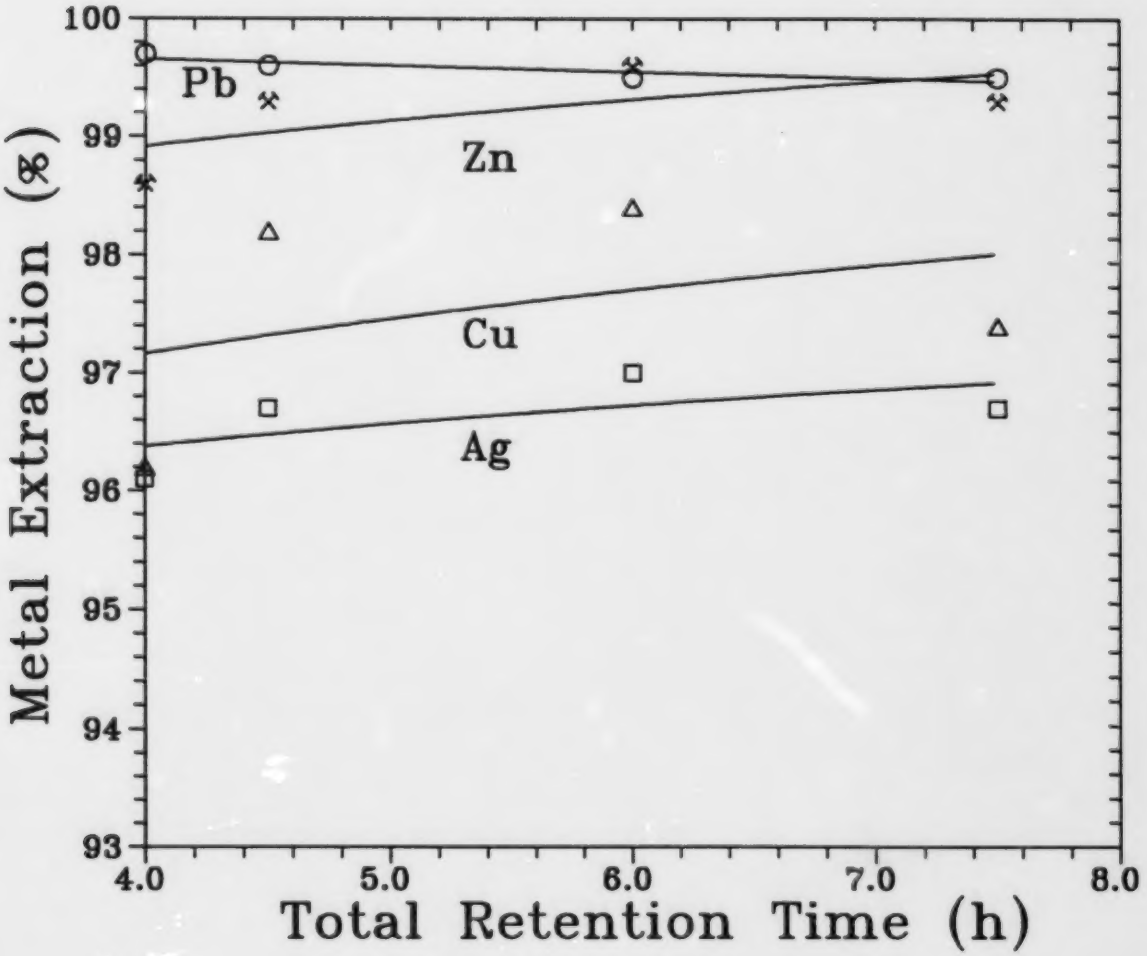

Figure 7 Cumulative Stage 1 and 2 Metal Extraction (0.10M HCl)